DESIGN OF SUPPORT SYSTEM FOR BORD AND PILLAR WORKINGS

A THESIS SUBMITTED IN PARTIAL FULFILLMENT
OF THE REQUIREMENTS FOR THE DEGREE OF

BACHELOR OF TECHNOLOGY
IN
MINING ENGINEERING

BY

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2010

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Under the Guidance of
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CERTIFICATE

This is to certify that the thesis entitled, "***Design of support system for bord and pillar workings***" submitted by Mr. Manoj Kumar Biswal in partial fulfillment of the requirement for the award of Bachelor of Technology Degree in Mining Engineering at the National Institute of Technology, Rourkela (Deemed University) is an authentic work carried out by him under my supervision and guidance.

To the best of my knowledge, the matter embodied in the thesis has not been submitted to any University/Institute for the award of any Degree or Diploma.

Date –

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ACKNOWLEDGEMENT

The most pleasant point of presenting a thesis is the opportunity to thank those who have contributed to it. Unfortunately, the list of expressions of thank no matter how extensive is always incomplete and inadequate. Indeed this page of acknowledgment shall never be able to touch the horizon of generosity of those who tendered their help to me.

First and foremost, I would like to express my gratitude and indebtedness to Dr. S. Jayanthu for his kindness in allowing me to do work in the present topic and for his inspiring guidance, constructive criticism and valuable suggestion throughout this project work. I am sincerely thankful to him for his able guidance and pain taking effort in improving my understanding of this project.

An assemblage of this nature could never have been attempted without reference to and inspiration from the works of others whose details are mentioned in reference section. I acknowledge my indebtedness to all of them.

And my sincere thanks to all my friends who have patiently extended all sorts of help for accomplishing this undertaking.

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ABSTRACT

The estimation of rock load from the strata and its distribution over the underground mine workings is of prime importance. In Indian coalmines, CMRI- RMR and NGI-Q Systems are mostly used for formulating design of support in rock engineering. Support systems are also designed with the help of numerical modeling. The working has been modeled by writing a program code in FLAC5.0. The modeling is done for driving of galleries (development) to form three pillars. The vertical stress contours indicated a maximum stress of about 2.5Mpa on pillars for the depth of 53m with a gallery width of 4.8 m and pillar size of 25 m.

Rock load on the basis of RMR for the **development gallery** is 2.4 t/m^2. Support system consisting of Resin bolt with bolt spacing of 1.5 m row spacing of 1.5 m has a support capacity of 5 t/m^2. No of bolts in a row is 3 is for galleries of 4.5m width and area of 4.5m x 1.5m. This support system has a safety factor of 2, which is adequate. Rock load on the basis of RMR for the junction is 3.17t/m^2. support system consisting of Resin bolt with bolt spacing of 1.5 m, row spacing of 1.5 m, has a support capacity of 7.4t/m^2 No of bolts in a row is 3, No of row 3, is proposed to support junction of 4.5m x 4.5m area which gives a safety factor of 2.33. It shows the junction is well supported & there is no fear of roof fall, so support is adequate.

For depillaring workings, Rock load on the basis of Q is 6.19 t/m^2 for Slice, and 7.79 t/m^2 for goaf edges. Support system consisting of one chock and 3 steel prop is designed. Goaf edge support is designed with skin to skin chock with a spacing of 1.2 with corner props. The support system for the slice consists of 2 props with spacing of 1.2m having a support capacity of 10 t/m^2. The support system gives a factor of safety of 1.61 for slices and safety factor of 1.23 for goaf edge.

Support system can be included in the numerical model for better understanding of the stability of the workings with supports of different types and capacities. These approaches can be followed for many mines and the performance of the support system can be monitored with load cells, and the models can be calibrated accordingly. This will improve the dependability and applicability of the numerical models as a solution for design of support system.

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CHAPTER 1

INTRODUCTION

The estimation of rock load from the strata and its distribution over the underground mine workings is of prime importance. In Indian coalmines, CMRI- RMR and NGI-Q Systems are mostly used for formulating design of support in rock engineering. In this project work their estimation is been done and design of support system done by CMRI – RMR & Q system. Support systems are also designed with the help of numerical modeling by simulating the workings. Study and analysis of the stress distribution around development workings in coal mines is conducted through numerical model.

1.1 Accident Statistics of Indian Underground Coal Mines

India has large resources of coal deposits for underground mining & lot of coal was blocked in existing underground mines. Safe extraction of these can be made possible by effective strata management & proper support design. Accident due to movement of strata in underground coal mines had been a major concern for the mining industry & its largest contributing factor of underground coal mine accident. Continuous efforts were being made by all concerns to reduce hazard of strata movement. To reduce strata movement monitoring of strata & proper design of support system is essential.

Table 1: Cause wise fatal accidents in coal mines, due to strata movement

Year	Fall of roof	Fall of sides	Total	Total B/G ACC	Percentage of accidents due to strata movement
1997	38	12	50	94	53
1998	35	15	50	80	62
1999	33	11	44	74	59.5
2000	27	14	41	62	66
2001	30	9	39	67	58
2002	23	11	34	48	70
2003	18	5	23	46	50
2004	26	8	34	49	69

2005	18	7	25	49	51
2006	13	4	17	44	40
2007	13	4	17	25	68
2008	13	7	20	33	60
Total	287	107	394	671	59.1

The analysis of accidents due to strata movement for last 12 years (1997- 2008) revealed that:

a) The roof fall & side fall accidents accounted for 59% of all below ground fatal accidents in coal mines.
b) Accidents due to fall of roof occurred in almost same proportion in bord & pillar development as well as depillaring districts.

The cause of the roof fall is due to improper design of support system. So in order to decrease the accident & increase productivity we need to design proper support system.

1.1.1 Objective

Objective of the present work is to design support system in development and depillaring workings by conventional method and numerical modeling of support system for development workings.

1.2 Present Support System

To design support, Systematic Support Rules (SSR) should be followed. Designing optimized support system means we have given proper support not less or not more, it helps in cost control of supports. The following are some of the design parameters:

i. Strata behavior
ii. Depth of cover
iii. Method of extraction
iv. Equipment selection for excavation
v. Span of area
vi. Height of extraction

The present support system for different working places in development and depillaring workings are as follows:

1.2.1 Goaf edges

At goaf edges cogs shall be set skin to skin. Props shall be set in between cogs, cogs & coal sides.

1.2.2 Working faces

1. At working faces props shall be set at a maximum interval of 1.2 m between the rows of props or in the same row.
2. Cogs shall be set at all entrances to the areas under extraction & also at interval of not more than 2.4m in the area under actual extraction.
3. Areas close to the faces where supports are likely to be affected due to blasting shall be supported by cross bars.

1.2.3 Support of galleries

- Props shall be set at interval of 1.2 m between them in the same row & at a max interval between rows of props in all galleries & splits within a distance of 2 pillars from the pillar under extraction or a distance of 30m, which ever greater.
- Cogs shall be set at all junctions.

1.2.4 Wider gallery

Gallery greater than 4.8 meter shall be supported with cogs at interval not exceeding 2.4m between cogs and between rows of cogs.

CHAPTER 2

LITERATURE REVIEW

Critical review of literature indicated the following approaches for estimation of rock load and design of support system for development and depillaring workings in underground coalmines.

2.1 Rock Load

Maximum load (P) that is required to be supported in the split and slice can be estimated using the following formula and as detailed elsewhere [Kushwaha, 2005]:

$$P = \gamma . SF1.5h \quad \text{………………..} \quad (1)$$

Where, γ = weighted average rock density, 2.5 t/m3 (carbonaceous shale)

SF1.5h = height of safety factor contour up to 1.5 in the roof strata in the simulated model.

2.2 Support Estimation

A pattern of support may be proposed using the following formula such that an adequate support safety factor (about 1.1-1.25 in depillaring areas, about 1.5-2.0 for permanent roadways) is achieved:

$$S = \frac{n \ x \ b_c}{w \ x \ sp} \quad \text{………………} \quad (2)$$

Where, n = the number of bolts/props in a row

bc= fully column grouted roof bolt capacity, 8 tonne fully column resin roof bolt capacity, 16 tonne capacity of timber props, 10 tonne capacity of timber cogs, 20 tonne

w =width of the slice, here 4.2m

sp= spacing between two rows

Support safety factor = **S / P**

2.3 Guidelines for Drawing of Support Plans in Bord and Pillar Workings in Coal Mines

The various stages of designing a suitable support system and ensuring successful installation are basically as follows:

(a) A geotechnical survey and interpretation of survey findings

(b) Selection/designing of support system based on above interpretation

(c) Selection of equipment

(d) Actual installation process and

(e) Monitoring of the system.

Two systems are particularly used to characterize mining ground conditions.

2.3.1 Barton's Q-system (Rock quality index, Norwegian Geotechnical Institute)

The rock quality index (Q) is evaluated as

$$Q = \frac{RQD \times j_r \times j_w}{j_n \times j_a \times SRF} \quad \ldots\ldots\ldots\ldots(3)$$

Where *RQD* = rock quality designation

Jn = joint set number

jr = joint roughness number

ja = joint alteration number

jw = joint water reduction number

SRF = stress reduction factor.

Based on the value of Q the rock mass can be described as "exceptionally good" (Q=400 to 1000) to "exceptionally poor" (Q=0.001 to 0.01). Using the Q value, the maximum unsupported span of roof can be estimated by the formula:

$$\text{Span (m)} = 2 \times ESR \times Q^{0.4} \ldots\ldots\ldots\ldots\ldots(4)$$

Where ESR is excavation support ratio (which is 3 to 5 for temporary mine workings and 1.6 for permanent workings). The rock load (Proof) can be estimated from the empirical formula:

$$\text{Proof (t/m2)} = \frac{2.0 * F}{Jr * Q\ 0.33} \quad \ldots\ldots\ldots\ldots\ldots(5)$$

Where F = 1 if Jr is 9 or more

Or F= ((Jr 0.5)/3 if Jr is less than 9.

Depending on the different values of the parameters and Q, 38 support categories have been identified.

2.3.2 Bieniawski's RMR system

There are five parameters in this classification:

(i) Intact rock strength

(ii) RQD

(iii) Joint spacing

(iv) Condition of joints

(v) Ground water seepage

Rating division for each of the parameters is given and RMR is sum of five ratings. Based on RMR, the roof is classified as very good (RMR:80-100) to very poor (RMR:0-20). From this estimation of rock load is derived using theoretical relation and support guide is provided.

These two classifications have been applied to about 30 Indian coal mines. The Q classification is suitable for highly jointed rocks for hard rock conditions. Most of the parameters in this system are based on joint attributes whereas stability in coal mines is not merely joint controlled. The SRF has no relation with the stress field occurring around multiple openings like coal mine roadways. The parameter descriptions in Q system leave much to subjective judgments.

The RMR system gives results nearer to actual roof conditions. it was recognized that in the most of the Indian coal mines, bedding planes, structural features and weathering of roof rocks are then major causes of roof failure. In Bieniawski's approach, consideration is not given to sedimentary features, structural features other than joints and weatherability of rocks. Deviations in the results also arise from the weightages for the parameters which need to be adjusted to Indian rock conditions.

2.4 CMRI-RMR Rock Mass Classification

This rock mass classification system is being used regularly by academic and research institutes. The five parameters used in the classification system and their relative ratings are summarized below:

Table 2. CMRI-RMR prescribed parameters for RMR determination

Sl No	Parameter	Max. rating
1	Layer thickness	30
2	Structural features	25
3	Rock weatherability	20
4	Strength of roof rock	15
5	Ground water seepage	10

The five parameters should be determined individually for all the rock types in the roof upto a height of at least 2 m.

A. Laying thickness: Spacing between the bedding planes or planes of discontinuities should be measured using borehole strata scope in an e m long drill made in the roof. Alternately, all bedding planes or weak planes within the roof strata can be measured in any roof exposure like a roof fault area, shaft section or cross measure drift. Core drilling shall be attempted wherever feasible and the core log can be used to evaluate RQD and layer thickness. Average of five values should be taken and layer thickness should be expressed in cm.

B. Structural Features: Random geological mapping should be carried out and all the geological features (discontinuities like joints, faults and slips, and sedimentary features like cross bedding, sandstone channels) should be carefully recorded. The relative orientation, spacing and degree of abundance for all these features shall be noted. Their influence on gallery stability should be assessed and the structural index for each feature should be determined from the Table 1 as given below.

C. Weatherability: ISRM standard slake durability test should be conducted on fresh samples from the mine to determine the susceptibility of rocks to weathering failure on contact with water or the atmospheric moisture. For this test, weigh exactly any ten irregular pieces of the sample (the total weight should be between 450- 500 g); place them in the test drum immersed in water and rotate it for 10 min at 20 rpm; dry the material retained in the drum after the test and weigh it again. Weight percentage of material remaining after test is the final slake durability index, expressed in percentage. Mean of three such first cycle values should be taken. Core may be broken to obtain the samples.

D. Rock Strength: Point load test is the standard index text for measuring the strength of rocks in the field. Irregular samples having ratio of 2:1 for longer axis to shorter axis can be sued for the test. The sample is kept between the pointed platens and the load is applied gently but steadily. The load at failure in kg divided by the square of the distance between the platens in cm gives the point load index (Is). The mean of the highest five values out of at least 10 sample tests should be taken. The compressive strength of the rocks can be obtained from the irregular lump point load index for Indian coal measure rocks by the relation: Co = 14 Is (in kg/cm2)

E. Ground water: A 2m long vertical borehole should be drilled in the immediate roof and the water seeping through the hole after half an hour should be collected in a measuring cylinder. The average of three values from three different holes should be taken and expressed in ml/minute.

Rock Mass Rating (RMR) is the sum of five parameter ratings. If there are more than one rock type in the roof, RMR is evaluated separately for each rock type and the combined RMR is obtained as:

$$\textbf{Combined RMR} = \frac{\Sigma\,\textbf{(RMR of each bed x bed thickness)}}{\Sigma\,\textbf{(Thickness of each bed)}} \qquad \textbf{...... (6)}$$

The RMR so obtained may be adjusted if necessary to take account for some special situations in the mine like depth, stress, method of work

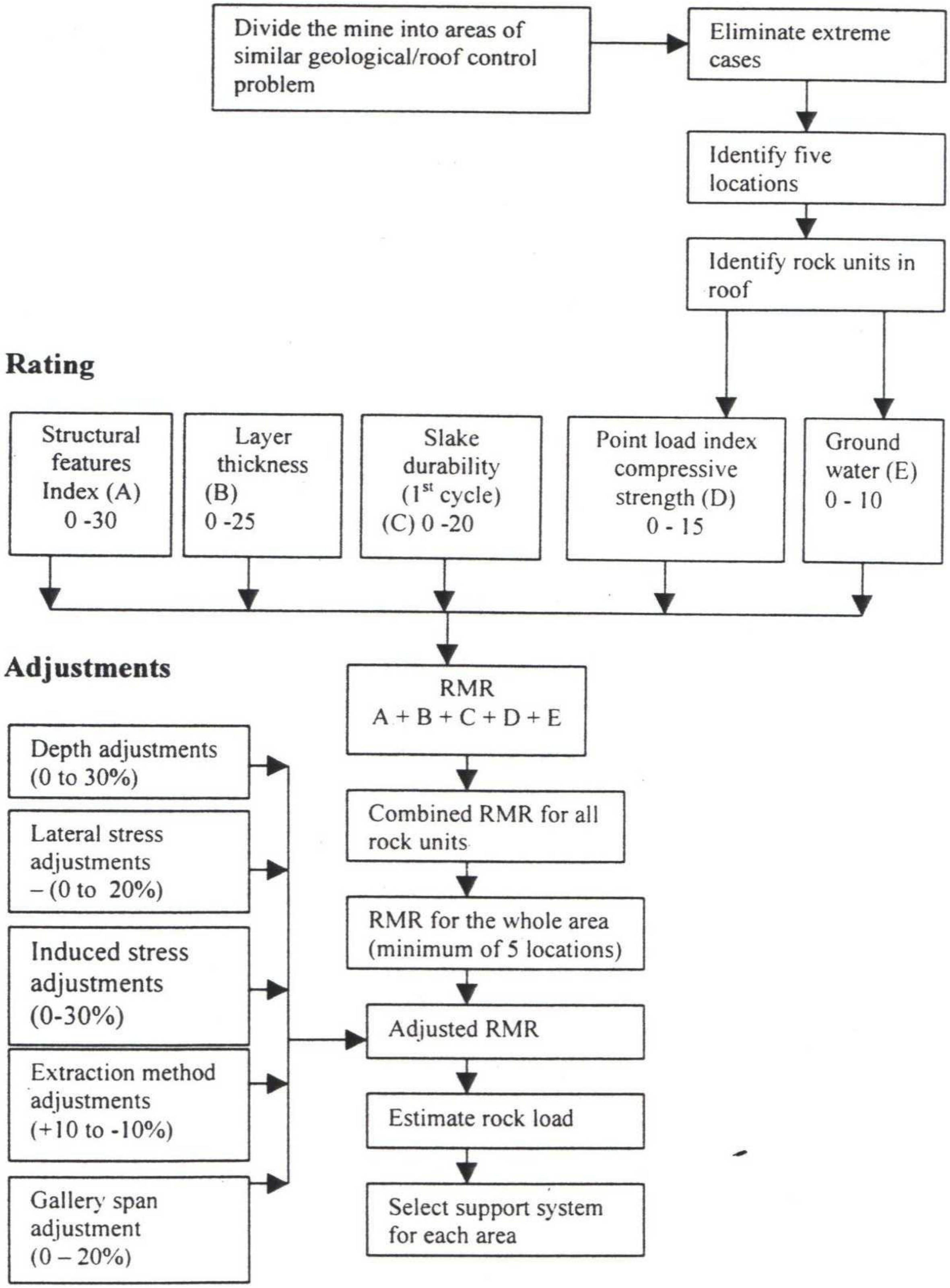

Figure .1 Flow-sheets for deriving RMR

Table 3: Classification of rock mass rating

Sl No	Rock mass rating	Rock quality
1	0 – 20	Very poor
2	20 – 40	Poor
3	40 – 60	Fair
4	60 – 80	Good
5	80 – 100	Very good

2.5 Design Of Support For Depillaring Working

In general, Rock Mass Rating (RMR) is used for design of supports in development galleries. However, due to limitations of its application to depillaring workings, many investigators adopted various approaches such as Q-classification of rock mass, numerical modeling etc for design of support system in depillaring workings, Sometimes, it is also required to design support in a depillaring panel having widely varying geo mining conditions with different support density.

For the purpose of support design in a typical depillaring area, Barton's Rock mass classification index- Q was also determined as follows:

$$\mathbf{Q = \{RQD/Jn\}\ \{Jr/Ja\}\ \{Jw/SRF\}\ \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ (7)}$$

Rock Quality Designation = f (layer thickness) = 97

Jn = no joints were observed in the roof = 4 for galleries

= 12 for junctions

= 20 for goaf edges

Ja = Plant impressions are frequent in the roof; however kettle bottoms/sandstone channels/slicken sides are not perceptible = 1

Jw = generally dry up to 8 ml of water per minute seepage.= 1

Jr = Smooth planar joints = 1

SRF values for various geometries during depillaring are as follows:

	C/Ms	SRF
For galleries and junctions:	>10	1
	1 - 10	1-2
For slices:	>5	2
	2.5 - 5	3 - 5
	<2.5	5
For goaf edges:	any value	10

Roof pressure could be estimated by the relations based on the Q value adjusted to the geometrical conditions:

For joint set number (Jn)> 9, the roof pressure (Proof) = 2/Jr x (5Q)-1/3

For Jn < 9, Proof = 2/3 Jn1/2 /Jr x (5Q)-1/3

2.6 Monitoring And Control of Strata Movement:

Strata and support behavior monitoring is required for understanding the performance of support system. Figure 2 illustrates the instrumentation required for strata monitoring in a typical development gallery of underground coal mine. To minimize the dangers from weighting on the pillar due to overhanging of roof in the goaf and to ensure that as small an area of un-collapsed roof as possible is allowed in the goaf, a suitable code of practice for induced blasting shall be evolved in consultation with a scientific organization keeping in view the depth of induce shot holes being not less than 2.7 m, direction & spacing of shotholes, explosives used etc. so as to limit the rate of convergence [i.e., the ratio of C1/C2 is equal or less than 2, where C1 is daily convergence at a site in a day "n" and C2 is the average daily convergence at the site up to the previous day i.e. day (n-1)] and also to ensure complete filling of the goaf and release of any abutment pressures.

Convergence recording stations shall be installed at all junctions situated within two pillar distance from pillar under extraction in the proposed panel. Monitoring of readings at convergence recording stations shall be done in every shift by a competent person duly authorized by the manager and the measurements shall be recorded in a bound paged book and the same shall be counter signed daily by the Under Manager of the shift and Asst. Manager in charge. All the work persons shall be withdrawn from the abutment zone if the ration of C1/C2 is equal to or more than 2 as given above and steps shall be taken to release the goaf abutment pressure by induced blasting. The Safety Officer shall co-ordinate recording, analysis and interpretation of the readings and advises the Officers/ Officials daily at the mine.

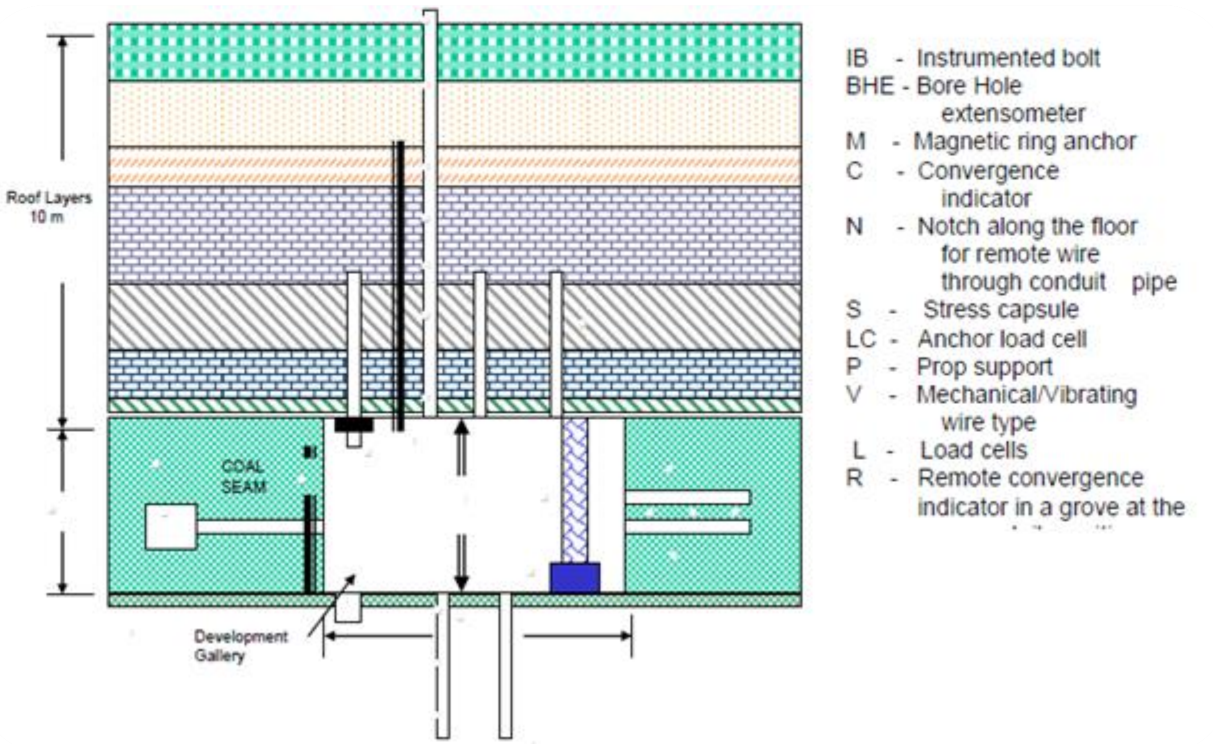

Figure .2 Typical Instruments for Strata Monitoring[Jayanthu et al 2008]

2.7 Numerical Modeling

A computer simulation, a computer model or a computational model is a computer program, or network of computers, that attempts to simulate an abstract model of a particular system. Models can take many forms, including but not limited to dynamical systems, statistical models, differential equations, or game theoretic models.

Often when engineers analyze a system to be controlled or optimized, they use a mathematical model. In analysis, engineers can build a descriptive model of the system as a hypothesis of how the system could work, or try to estimate how an unforeseeable event could affect the system. Similarly, in control of a system, engineers can try out different control approaches in simulations.

A mathematical model usually describes a system by a set of variables and a set of equations that establish relationships between the variables. The values of the variables can be practically anything; real or integer numbers, boolean values or strings, for example. The variables represent some properties of the system, for example, measured system outputs often in the form of signals, timing data, contours, and event occurrence (yes/no). The actual model is the set of functions that describe the relations between the different variables. Here FLAC (Fast Lagrangian Analysis of Continua)5.0 has been used for simulation and analysis.

2.7.1 FLAC 5.0

FLAC is a two-dimensional explicit finite difference program for engineering mechanics computation. This program simulates the behavior of structures built of soil, rock or other materials that may undergo plastic flow when their yield limits are reached. Materials are represented by elements, or zones, which form a grid that is adjusted by the user to fit the shape of the object to be modeled. Each element behaves according to a prescribed linear or nonlinear stress/strain law in response to the applied forces or boundary restraints. The material can yield and flow and the grid can deform (in large-strain mode) and move with the material that is represented. The explicit, Lagrangian calculation scheme and the mixed-discretization zoning technique used in *FLAC* ensure that plastic collapse and flow are modeled very accurately. Because no matrices are formed, large two-dimensional calculations can be made without excessive memory requirements. The drawbacks of the explicit formulation (i.e., small timestep limitation and the question of required damping) are overcome to some extent by automatic inertia scaling and automatic damping that do not influence the mode of failure.

Though *FLAC* was originally developed for geotechnical and mining engineers, the program offers a wide range of capabilities to solve complex problems in mechanics. Several built-in constitutive models that permit the simulation of highly nonlinear, irreversible response representative of geologic, or similar, materials are available.

However, it offers several advantages when applied to engineering problems.

a. The input language is based upon recognizable word commands that allow you to identify the application of each command easily and in a logical fashion (e.g., the **APPLY** command applies boundary conditions to the model).
b. Engineering simulations usually consist of a lengthy sequence of operations — e.g., establish in-situ stress, apply loads, excavate tunnel, install support, and so on. A series of input commands (from a file or from the keyboard) corresponds closely with the physical sequence that it represents.
c. A *FLAC* data file can easily be modified with a text editor. Several data files can be linked to run a number of *FLAC* analyses in sequence. This is ideal for performing parameter sensitivity studies.
d. The word-oriented input files provide an excellent means to keep a documented record of the analyses performed for an engineering study. Often, it is convenient to include these files as an appendix to the engineering report for the purpose of quality assurance.
e. The command-driven structure allows you to develop pre- and post-processing programs to manipulate *FLAC* input/output as desired. For example, you may wish to write a mesh generation function to create a special grid shape for a series of *FLAC* simulations. This can readily be accomplished with the *FISH* programming language, and incorporated directly in the input data file

CHAPTER 3

DESIGN OF SUPPORT SYSTEM

For design of support, rock load is estimated, and various support types are considered for the geo mining condition of the site. Capacity for various support systems tested in a premier research institute (Central Institute for Mining and Fuel Research, Dhanbad) is widely adopted by the investigators. Table4 gives the support capacity for different type of supports such a props, chocks, bolts etc.

Table -: 4 Types of supports used in coal mines & their load bearing capacity (CMRI Report, 1987)

Sl No	Support item	Load bearing capacity (t)
1	Roof bolt (full column grouted with quick setting cement capsules) (TMT ribbed bolt of 22 mm diameter)	6
2	Roof bolt (full column grouted with resin capsules) (TMT ribbed bolt of 22 mm diameter)	12
3	Roof stitching	8
4	Wooden prop	10
5	Steel prop	30
6	Steel chock	30
7	Wooden chock	20
8	Pit prop	15

Table 5: RMR calculation of given data

Sl No	**Parameter**	**Description**	**Rating**
1	Layer thickness	17cm	19
2	Structural features	Joint slip (indices = 8)	14
3	Weatherability	91%	11
4	Compressive strength	215 kg/cm^2	05
5	Ground water	Moist	09
	Total RMR		58 (fair rock)

Table 6: calculation of Q

						SRF		Q	
Parameters (→)	**RQD**	**Jn**	**Jr**	**Ja**	**Jw**	**slices**	**Goaf edges**	**slices**	**Goaf edges**
Value (→)	60	9	1.5	1	1	5	10	2	1

3.1 Geo mining Details

Seam thickness : 7 meters

Pillar size : 20m * 20m (centre to centre)

Gallery size : 4.5 m * 3m

Depth of cover : 100 meters

RMR : 58 (FAIR)

Slice width : 5m

3.2 Equations for Support Load

$$\textbf{\textit{Rock load}} = \boldsymbol{B \times D * F\,(1.7 - 0.037 \times RMR + 0.0002 \times RMR)^{2}} \ldots\ldots\ldots\ldots \textbf{(8)}$$

Where, B (width of galleries/splits),

D (average rock density)

F is safety factor

A safety factor of 1.5 is generally considered enough.

3.3 Estimation of Rock Load & Design of Support System in Development Working

3.3.1 Rock load in galleries and splits

Rock load (t/m^2) in the galleries and splits in depillaring areas has been determined using the empirical relationship of CMRI-RMR System i.e in (equation 9):

$$\textbf{\textit{Rock load}} = \boldsymbol{B \times D*F\,(1.7-0.037 \times RMR + 0.0002 \times RMR)^2} \ldots\ldots\ldots\ldots\ \mathbf{(9)}$$

RMR = 58

Gallery span (B, width) = 4.5 m

Density (D) = 2.29 t/m^2

Height = 3m

Safety factor (F) = 2 (generally)

Rock load = 4.80 t/m^2

Hence, rock load in galleries and splits = 4.80 t/m^2

To support the rock load we need to give support higher than the rock load at that area so that the roof will not fall. Type and capacity of various supports are chosen from Table 4 for design of Support system in bord and pillar workings. It is assumed that full column resin bolt is used as the support system.

Roof bolt capacity = 12 t

Bolt spacing = 1.5 m

Distance between two rows of bolts = 1.5 m

No of bolts in a row= 3

The support design shown in the Figure 3

$$\textbf{Applied support load} = \frac{\mathbf{(n * A) + (m * Q)}}{\mathbf{W * a}} \ldots\ldots\ldots\ldots\ldots\ldots\ \mathbf{(10)}$$

Where, n is number of bolts in a row

A is the anchorage strength of each bolt (t)

Q is the load bearing capacity of the additional support if done (t)

m is the number of additional support at spacing "a" if it has been used

W width of split or slice

a is the spacing between two consecutive rows

The above configuration leads to support resistance of

$$= (12 * 3) \text{ t} / (4.5 * 1.5) \text{ m}^2$$

$$= 5 \text{ t/m}^2$$

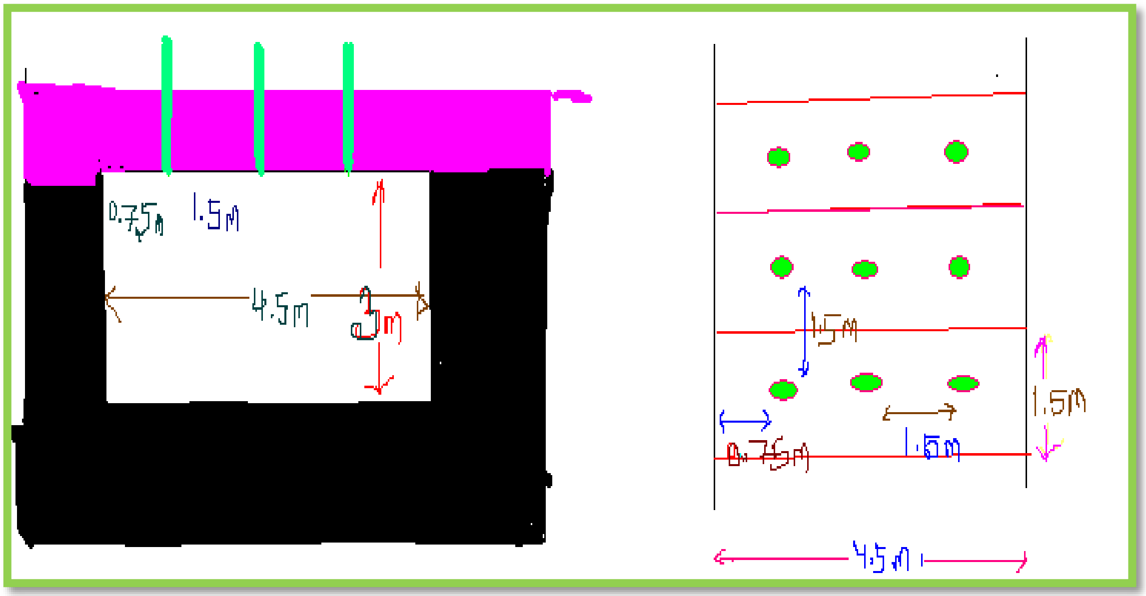

Figure 3: Support design for galleries

3.3.2 Rock load at junctions

Rock load at junction of gallery and split in depillaring areas has been estimated using the following empirical equation of CMRI-RMR system:

$$\textbf{\textit{Rock load}} = 5 \times B^{0.3} \times D\,(1\text{-}RMR/100)^2 \text{.......... } \mathbf{(11)}$$

Rock load $= 3.17$ t/m^2

To support the rock load we need to give support higher than the rock load at that area so that the roof will not fall. It's assumed that junctions will be supported by cement grouted bolt (6 ton) & chocks

Junction dimension = 4.5 * 4.5 m

Roof bolt capacity = 6 t

Bolt spacing = 1.5 m

Distance between two rows of bolts = 1.5 m

Bolts in middle row = 3 & in another two row of support chocks with 1 bolt in between.

So total bolt =5 & chock = 4 as given in the **figure4**

The above configuration leads to support resistance of **{(4*30) + (5*6)} / (4.5*4.5)**

$$\mathbf{=150\ t/20.25m^2 = 7.4\ t/m^2}$$

Safety factor = 7.4/3.17 = 2.33

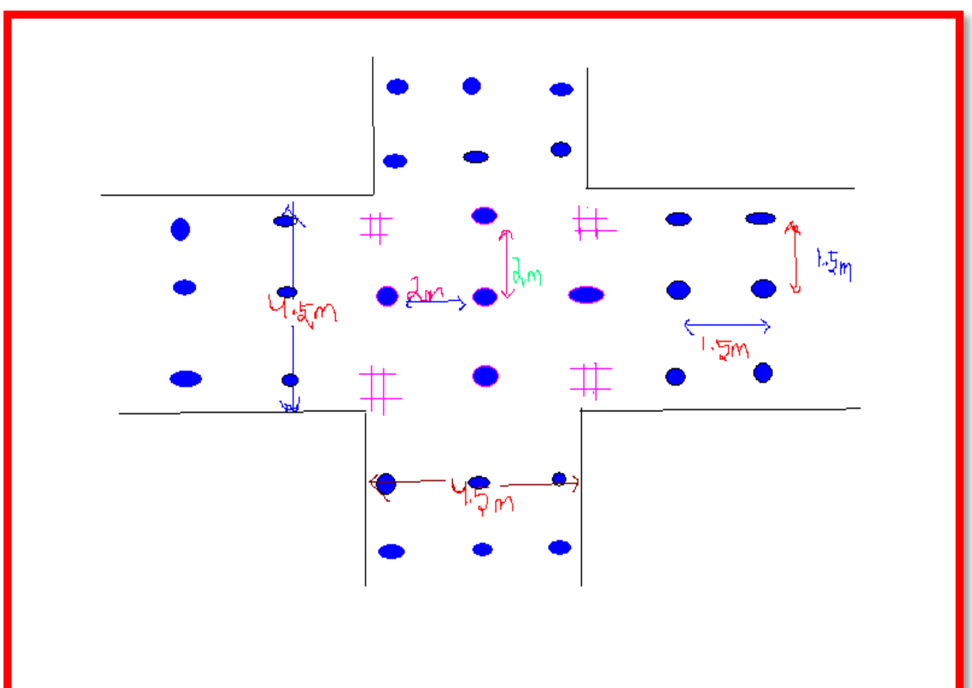

Figure 4: Support design for junctions

Another approach

It's assumed that junctions will be supported by full column resin bolt

Roof bolt capacity = 12 t

Bolt spacing = 1.5 m

Distance between two rows of bolts = 1.5 m

No of bolts in a row= 3

No of row = 3

The design shown in Figure 5

The above configuration leads to support resistance of **108 t/20.25sq.m = 5.33 t/sq.m**

Safety factor **= 5.33/3.17**

= 1.68

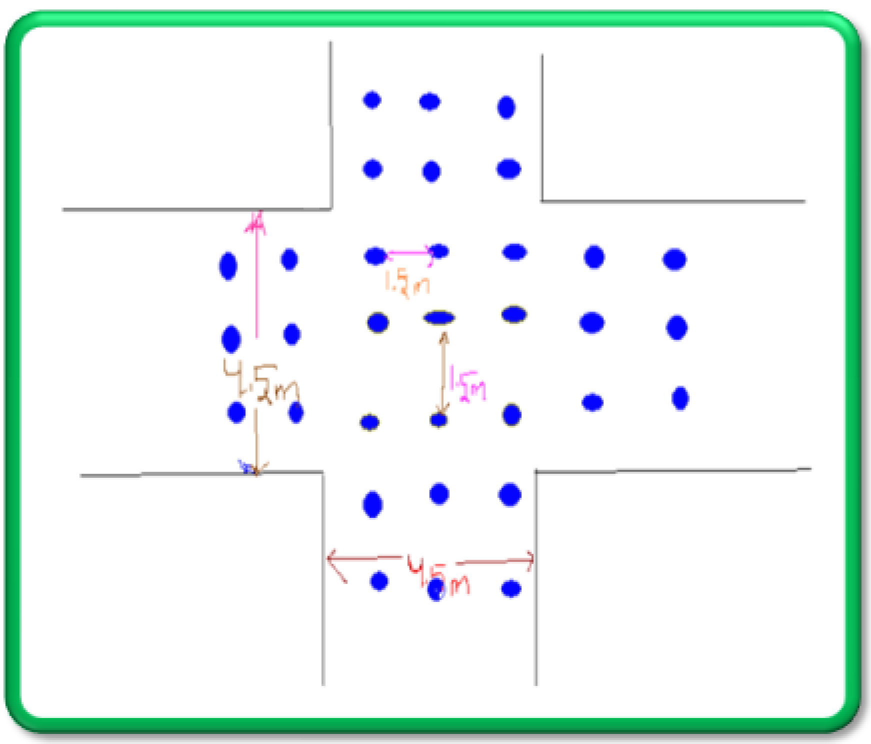

Figure 5: design of support system for junctions

3.4 Estimation of Rock Load & Design of Support System In (depillaring workings)

3.4.1 Rock load in slice and goaf edge

Rock load (P_{roof}) in slice and goaf edge was estimated using NGI-Q system from the following empirical relation:

$$Proof = 2/3\ (J_n^{1/2} / J_r)\ x\ (5Q)^{-1/3} \ldots\ldots\ldots\ldots\ldots (12)$$

Where, Jn = 9, Jr = 1.5, Q = 2 for slice and Q = 1 for goaf edge.
Hence, rock load in slice, *Proof* is 6.19 t/m^2; and rock load at goaf edge, *Proof* is 7.79 t/m^2. It's assumed that slice & goaf edges will be supported by steel props & chocks.

Slice width = 5m

Rock load in slice, *Proof* is 6.19 t/m^2

Steel prop capacity = 30 t

Prop spacing = 1.5 m

Distance between two rows of prop = 1.2 m

Chock capacity = 30 ton

In 2.4m of length support system will be 1 chock & 3 steel prop as shown in figure. Goaf edge side will have 1 chock & 1 prop with spacing 1.2 m. the other side will have 2 props with spacing 1.2m.

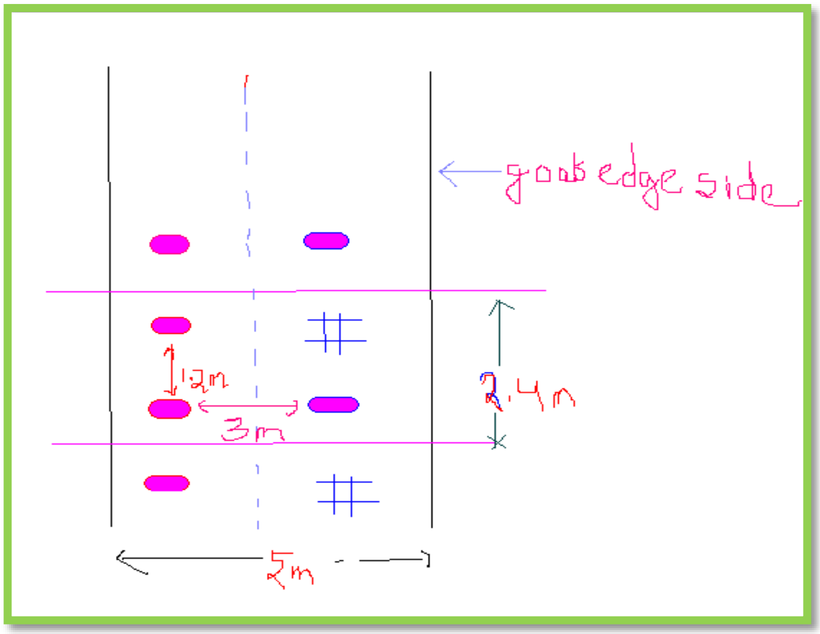

Figure 6: Design of support system for slices & goaf edges

The above configuration leads to support resistance of **120 t/12sq.m = 10 t/sq.m**

Safety factor = 10/6.19

= 1.61 for slices

For goaf edges one chock & 1 prop are there in 2.5m width & 2.4m length so

The support resistance is **= (30 + 30) / (2.5*2.4)**

= 60 / 6 = 10t/m^2

Safety factor = 10 / rock load for goaf edge

= 10 /7.79

= 1.28

It is good because goaf edge is supported for temporary period.

3.5 Numerical Modeling Study

The 3 pillars have been modeled using FLAC5.0 with 4 galleries. At galleries three cable bolts are provided for supporting the roof strata.

The modeling procedure is

a. OPEN FLAC 5.0
b. Go to GRID & make grid
c. Then go to ASSIGN to make galleries & give rock properties
d. Go to STRUCTURE to support gallery with rock bolt or cable bolt or props.
e. Then go to run for calculation of FOS
f. Go to PLOT to see different profiles

The galleries are supported by 3 bolts for each gallery and then its vertical stress contours are plotted. Program code for numerical model is given in Appendix 1. Descretisation of numerical model showing four galleries is presented in figure 7:

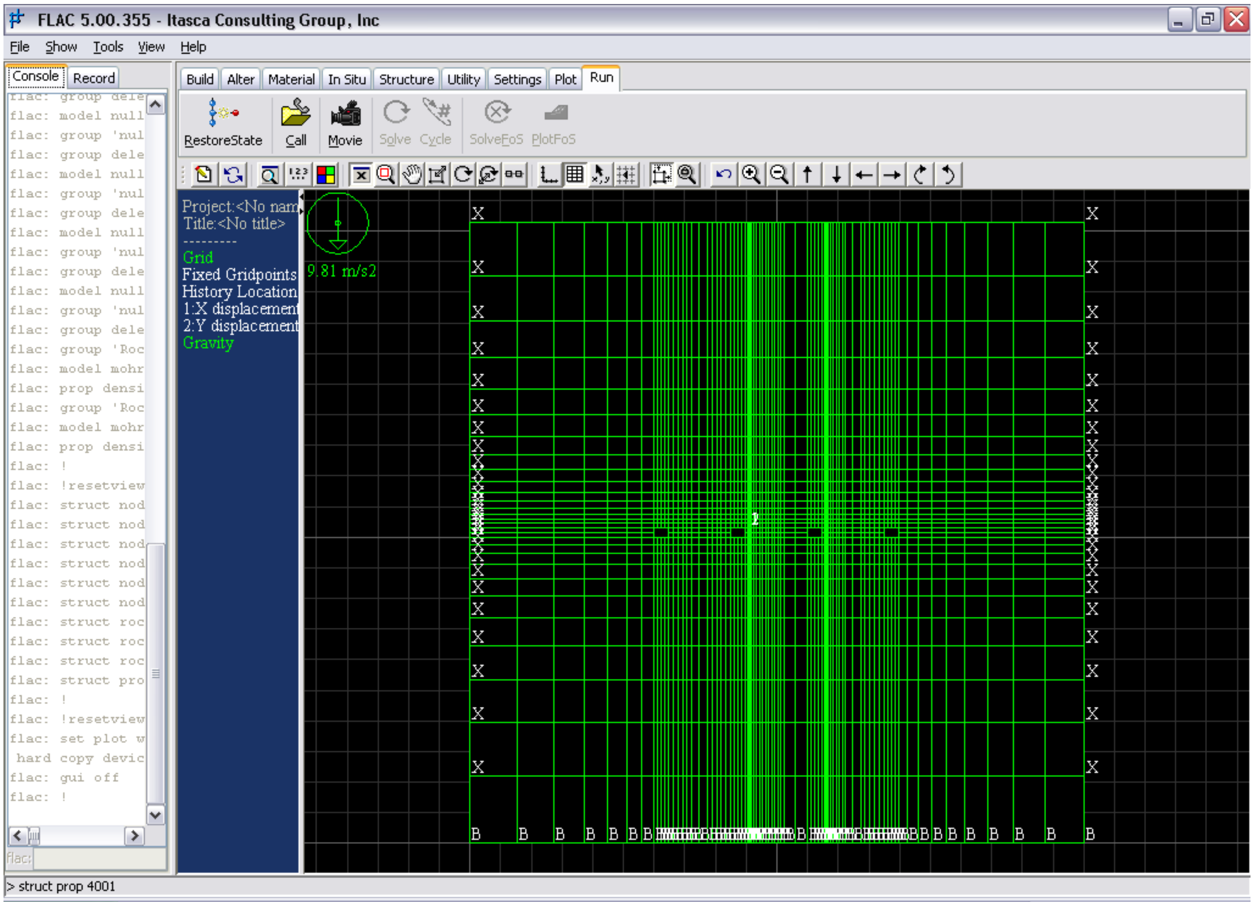

Figure 7: Descretisation of numerical model showing four galleries

CHAPTER 4

ANALYSIS AND RESULTS

4.1 Conventional System

Rock load or load from roof of development workings (gallery, junction) and in depillaring workings (slices, & goaf edges) was calculated by CMRI – RMR and Q system, respectively. Analysis of rock load was conducted and accordingly support system was designed.

The support system designed for development workings is discussed below:

a. For supporting gallery of 4.5 m width, i.e. for an area of 4.5m * 1.5m the rock load calculated to be 4.80t/m^2. Support systems of three bolts at 1.5 m spacing and interval between the rows of 1.5m are designed as shown in the figure3. Factor of safety calculated and it is 2. From the factor of safety we can conclude that the support is adequate for the gallery.
b. For supporting junctions of 4.5m * 4.5m the rock load calculated to be 3.17t/m^2. Support system of three Resin bolt capacity of 12 ton in a row at spacing of 1.5 m, and interval between the rows is of 1.5m, number of rows provided is 3 as shown in figure 4. Factor of safety calculated and is 1.68. It shows the junction is well supported & there is no fear of roof fall.

The support system designed for depillaring workings including slices and goaf edges is as follows:

Since the slices are subjected to assymetrical loading due to goaf on one side and stook on the other side, support system should also be assymetrical with more towards the goaf side as compared to other side of the slice. There fore, alternate chock and prop are generally used on goaf side where as the other side is supported with a row of props. Goaf edges in a depillaring panel should be fenced to eliminate the chance of any body entering in to the goaf. Thus, the objective of goaf edge support is not only to give resistance to fall of roof beyond the goaf line but also to ensure that employees are kept safe from the fallen goaf material.

Rock load on the basis of Q is 6.19 t/m^2 for Slice, and 7.79 t/m^2 for goaf edges. Support system consisting of one chock and 3 steel prop is designed. Goaf edge support is designed with skin to skin chock with a spacing of 1.2 with corner props. The support system for the slice consists of 2 props with spacing of 1.2m having a support capacity of 10 t/m^2. The support system gives a factor of safety of 1.61 for slices and safety factor of 1.23 for goaf edge.

4.2 Numerical Modeling by FLAC 5.0

Simulation of bord and pillar workings at a depth of 53m was carried out through numerical model. Stress distribution around the development workings is shown in Figure 9. The 3 pillars have been modeled using FLAC5.0 with 4 galleries. The particulars about the seam which has been modeled are it has Seam thickness of 7.5m, Pillar size of 25m, Depth of cover of 53m, the Gallery size is of 4.8m X 3m, split Width of 5m ; Rib thickness 2.5m The stress is more at the corner of the pillar i.e. 2.5Mpa for 53m of depth. There is no formation of tension zone, so the 3 bolts are adequate to support a gallery.

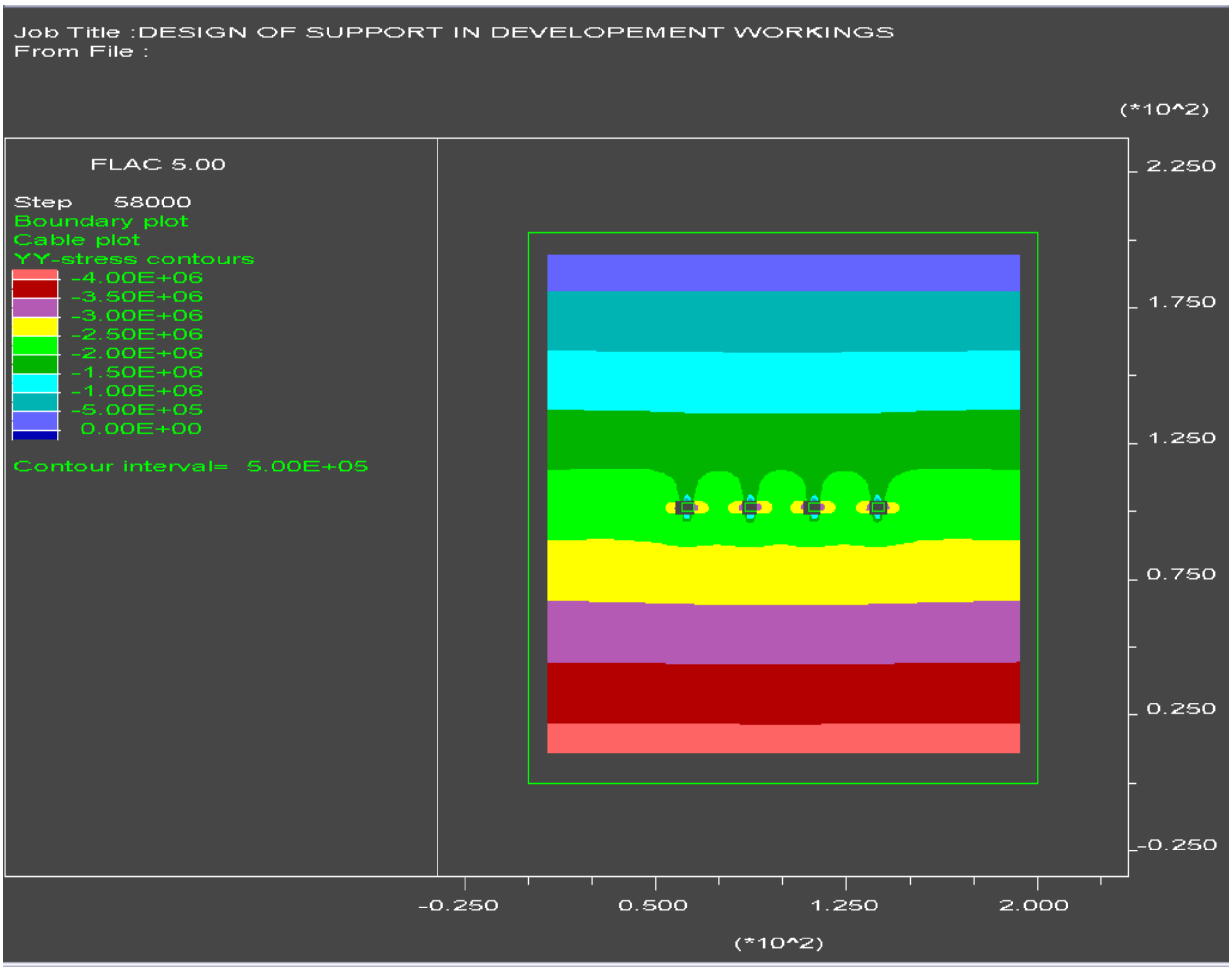

Figure 8: stress distribution around galleries

CHAPTER 5

CONCLUSIONS AND RECOMMENDATIONS

Based on the critical review and application of various approaches for designing of support system for development and depillaring workings of underground coal mines with a case study, the following are the conclusions and recommendations:

CONCLUSIONS:

1. for development workings

1.1 Rock load on the basis of RMR for the development gallery is 2.4 t/m^2. Support system consisting of Resin bolt with bolt spacing of 1.5 m row spacing of 1.5 m has a support capacity of 5 t/m^2. No of bolts in a row is 3 is for galleries of 4.5m width and area of 4.5m x 1.5m. This support system has a safety factor of 2, which is adequate.

1.2 Rock load on the basis of RMR for the junction is 3.17t/m^2. support system consisting of Resin bolt with bolt spacing of 1.5 m, row spacing of 1.5 m, has a support capacity of 7.4t/m^2 No of bolts in a row is 3, No of row 3, is proposed to support junction of 4.5m x 4.5m area which gives a safety factor of 2.33. It shows the junction is well supported & there is no fear of roof fall, so support is adequate.

1.3 modeling by FLAC 5.0 done with 3 pillars and stress contour plotted from which it is found that The stress is more at the corner of the pillar i.e. 2.5Mpa for 53m of depth of cover. There is no formation of tension zone, so the 3 cable bolts are sufficient to support a gallery.

2. for depillaring workings

Rock load on the basis of Q is 6.19 t/m^2 for Slice, and 7.79 t/m^2 for goaf edges. Support system consisting of one chock and 3 steel prop is designed. Goaf edge support is designed with skin to skin chock with a spacing of 1.2 with corner props. The support system for the slice consists of 2 props with spacing of 1.2m having a support capacity of 10 t/m^2. The support system gives a factor of safety of 1.61 for slices and safety factor of 1.23 for goaf edge.

RECOMMENDATIONS

1. Support system can be included in the numerical model for better understanding of the stability of the workings with supports of different types and capacities.

2. These approaches can be followed for many mines and the performance of the support system can be monitored with load cells, and the models can be calibrated accordingly. This will improve the dependability and applicability of the numerical models as a solution for design of support system.

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(2) *D. P. Tripathy, 2008* CMRI-ISM Rock Mass Classification, Short Term Course on "Trends in Strata Control Techniques & Instrumentation for Enhancing Safety in Coal Mines" July 28 – 31, 2008 Mining Dept. NIT Rourkela (page 87-97)

(3) Ajoy K Singh, A Sinha, A Paul and K Saikia, 2005 NGI- Q system and CMRI-RMR classification, paper on "Geotechnical investigation for support design in depillaring panels in Indian coalmines" Journal of Scientific & Industrial Research Vol. 64, May 2005, pp. (358-363)

(4) *S Jayanthu, V Venkateswarlu, 2008* Strata Behahiour in Development and Depillaring workings supported with Roof Bolt and Cable Bolts, Short Term Course on "Trends in Strata Control Techniques & Instrumentation for Enhancing Safety in Coal Mines" July 28 – 31, 2008 Mining Dept. NIT Rourkela (page 145-153)

(5) A Kushwaha et.al 2006 "Development of support guidelines for depillaring panels in Indian coal mines" CIMFR Report, October 2001 to December 2006 Central Institute of Mining and Fuel Research (page 45- 48)

(6) http://nopr.niscair.res.in/bitstream/123456789/5151/1/JSIR%2064(5)%20358-363.pdf

(7) http://www.ieindia.org/pdf/87/a06m135.pdf

(8) http://www.scribd.com/doc/30734336/Reduction-of-risk-from-roof-and-side-fall-in-Indian-coal-mines

APPENDIX – I

PROGRAM CODE FOR SIMULATION OF DESIGN OF SUPPORT SYSTEM OF DEVELOPMENT WORKINGS

```
TITLE
DESIGN OF SUPPORT IN DEVELOPEMENT WORKINGS
*PROGRAM DEVELOPED UNDER THE GUIDANCE OF PROF. S JAYANTHU
* Seam thickness=7.5m ( Panel 16), Pillar size=25m, Depth=53m
* Gallery size=4.8m X 3m, Width of split=5m ; Rib thickness=2.5m
*PARAMETRIC STUDIES
* Seam thickness=3-11m @2m, Pillar size=25m, Depth=30-240m @30m
* Gallery size=4.8m X 3m, Width of split=5m ; Rib thickness=2.5m
GR 78 29
M M
*
* Floor of the seam no 3 -100m
gen 0,0 0,100 60,100 60,0  R .8 .8                          I 1 8   J 1 12
gen 60,0 60,100 64.8,100 64.8,0 R 1 .8                      I 8 12  J 1 12
gen 64.8,0 64.8,100 72.25,100 72.25,0 R 1 .8                I 12 17 J 1 12
gen 72.25,0 72.25,100 77.25,100 77.25,0 R 1 .8              I 17 19 J 1 12
gen 77.25,0 77.25,100 85,100 85,0 R 1 .8                    I 19 24 J 1 12
gen 85,0 85,100 89.8,100 89.8,0 R 1 .8                      I 24 28 J 1 12
gen 89.8,0 89.8,100 92.3,100 92.3,0  R 1 .8                 I 28 33 J 1 12
gen 92.3,0 92.3,100 97.25,100 97.25,0  R 1 .8               I 33 38 J 1 12
gen 97.25,0 97.25,100 102.25,100 102.25,0 R 1 .8            I 38 43 J 1 12
gen 102.25,0 102.25,100 110,100 110,0 R 1 .8                I 43 45 J 1 12
gen 110,0 110,100 114.8,100 114.8,0 R 1 .8                  I 45 49 J 1 12
gen 114.8,0 114.8,100 117.3,100 117.3,0 R 1 .8              I 49 54 J 1 12
gen 117.3,0 117.3,100 122.25,100 122.25,0 R 1 .8            I 54 59 J 1 12
gen 122.25,0 122.25,100 127.25,100 127.25,0 R 1 .8          I 59 61 J 1 12
```

```
gen 127.25,0 127.25,100 135,100 135,0  R 1 .8                I 61 66 J 1 12
gen 135,0 135,100 139.8,100 139.8,0  R 1 .8                  I 66 70 J 1 12
gen 139.8,0 139.8,100 200,100 200,0   R 1.2 .8               I 70 79 J 1 12
*
*Coal seam -7.5m
gen 0,100 0,107.5 60,107.5 60,100                    R .8 1 I 1 8    J 12 17
gen 60,100 60,107.5 64.8,107.5 64.8,100              R 1 1  I 8 12   J 12 17
gen 64.8,100 64.8,107.5 72.25,107.5 72.25,100               R 1 1  I 12 17  J 12 17
gen 72.25,100 72.25,107.5 77.25,107.5 77.25,100             R 1 1  I 17 19  J 12 17
gen 77.25,100 77.25,107.5 85,107.5 85,100            R 1 1  I 19 24  J 12 17
gen 85,100 85,107.5 89.8,107.5 89.8,100              R 1 1  I 24 28  J 12 17
gen 89.8,100 89.8,107.5 92.3,107.5 92.3,100          R 1 1  I 28 33  J 12 17
gen 92.3,100 92.3,107.5 97.25,107.5 97.25,100               R 1 1  I 33 38  J 12 17
gen 97.25,100 97.25,107.5 102.25,107.5 102.25,100      R 1 1  I 38 43  J 12 17
gen 102.25,100 102.25,107.5 110,107.5 110,100               R 1 1  I 43 45  J 12 17
gen 110,100 110,107.5 114.8,107.5 114.8,100                 R 1 1  I 45 49  J 12 17
gen 114.8,100 114.8,107.5 117.3,107.5 117.3,100             R 1 1  I 49 54  J 12 17
gen 117.3,100 117.3,107.5 122.25,107.5 122.25,100      R 1 1  I 54 59  J 12 17
gen 122.25,100 122.25,107.5 127.25,107.5 127.25,100  R 1 1  I 59 61  J 12 17
gen 127.25,100 127.25,107.5 135,107.5 135,100               R 1 1  I 61 66  J 12 17
gen 135,100 135,107.5 139.8,107.5 139.8,100                 R 1 1  I 66 70  J 12 17
gen 139.8,100 139.8,107.5 200,107.5 200,100                 R 1.2 1 I 70 79 J 12 17
*
*XXxxGraphite band  - 10 cm thick
* Sandstone roof-153m
gen 0,107.5 0,203 60,203 60,107.5                    R .8 1.2 I 1 8    J 17 30
gen 60,107.5 60,203 64.8,203 64.8,107.5                     R 1 1.2  I 8 12   J 17 30
gen 64.8,107.5 64.8,203 72.25,203 72.25,107.5        R 1 1.2  I 12 17  J 17 30
gen 72.25,107.5 72.25,203 77.25,203 77.25,107.5      R 1 1.2  I 17 19  J 17 30
gen 77.25,107.5 77.25,203 85,203 85,107.5            R 1 1.2  I 19 24  J 17 30
gen 85,107.5 85,203 89.8,203 89.8,107.5                     R 1 1.2  I 24 28  J 17 30
```

```
gen 89.8,107.5 89.8,203 92.3,203 92.3,107.5            R 1 1.2  I 28 33  J 17 30
gen 92.3,107.5 92.3,203 97.25,203 97.25,107.5         R 1 1.2  I 33 38  J 17 30
gen 97.25,107.5 97.25,203 102.25,203 102.25,107.5            R 1 1.2  I 38 43  J 17 30
gen 102.25,107.5 102.25,203 110,203 110,107.5       R 1 1.2  I 43 45  J 17 30
gen 110,107.50 110,203 114.8,203 114.8,107.5        R 1 1.2  I 45 49  J 17 30
gen 114.8,107.5 114.8,203 117.3,203 117.3,107.5     R 1 1.2  I 49 54  J 17 30
gen 117.3,107.5 117.3,203 122.25,203 122.25,107.5            R 1 1.2    I 54 59  J 17 30
gen 122.25,107.5 122.25,203 127.25,203 127.25,107.5  R 1 1.2 I 59 61  J 17 30
gen 127.25,107.5 127.25,203 135,203 135,107.5       R 1 1.2    I 61 66  J 17 30
gen 135,107.5 135,203 139.8,203 139.8,107.5          R 1 1.2    I 66 70  J 17 30
gen 139.8,107.5 139.8,203 200,203 200,107.5          R 1.2 1.2  I 70 79  J 17 30
PROP S=4.E9  B=6.67E9  D=2300 T=9.E6   C= 12.E6                  FRIC=45 I 1 78 J 1 11
PROP S=4.E9  B=6.67E9  D=2300 T=9.E6   C=12.E6                   FRIC=45 I 1 78 J 20 29
PROP S=2.2E9 B=3.67E9  D=1427 T=1.86E6 C=1.85E6    FRIC=30 I 1 78 J 12 16
PROP S=1.14E9 B=1.7E9    D=1850 T=.56E6 C=1.1E6                FRIC=35 I 1 78 J 17
PROP S=3.06E9 B=3.9E9  D=1850 T=2.8E6  C=2.1E6                 FRIC=35 I 1 78 J 19
PROP S=4.E9   B=6.67E9 D=2300 T=9.E6   C=12.E6                 FRIC=45 I 1 78 J 18
SET GRA 9.81
set large
FIX X  I 1
FIX X  J 1
FIX X  I 79
FIX Y  J 1
INI SYY -3.75E6 VAR 0 3.75E6
INI SXX -4.5E6 VAR 0 0.850E6
HIS NSTEP 10
HIS XDIS I 30 J 14
HIS YDIS I 30 J 14
*Development galleries 4.8m x 3m
HIS UNBAL I 1 J 1
*******************OPENING OF GALLERY 1***********
```

```
MOD NULL I  8 11  J 12 13
*******************OPENING OF GALLERY 2***********
MOD NULL i 24 27 j 12 13
*******************OPENING OF GALLERY 3***********
MOD NULL i 45 48 j 12 13
*******************OPENING OF GALLERY 4***********
MOD NULL i 66 69 j 12 13
RET
*SOLVE
**********************************************
*With developement only* Save as ncdev.sav
**********************************************
*Save  ncdev.sav
******Split galleries 5m x 3m
******************OPENING OF SPLIT 1**********
*MOD NULL I 17 18 J 12 13
*******************OPENING OF SPLIT 2**********
*MOD NULL i 38 42 j 12 13
*******************OPENING OF SPLIT 3**********
*MOD NULL i 59 60 j 12 13
* Cable bolts in the splits and galleries up to 2 pillar ahead of working
***********Gallery 1
STRUCT CABL BEG 61.2 103.   END 61.2 104.5.            PROP 1
STRUCT CABL BEG 62.4  103.  END 62.4  104.5.           PROP 1
STRUCT CABL BEG 63.6 103.   END 63.6 104.5.            PROP 1
STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9  A 3.14E-4
***********Gallery 2
STRUCT CABL BEG 86.2 103.   END 86.2 104.5.  PROP 1
STRUCT CABL BEG 87.4  103.  END 87.4  104.5. PROP 1
STRUCT CABL BEG 88.6 103.   END 88.6 104.5.            PROP 1
***********Gallery 3
```

```
STRUCT CABL BEG 111.2 103.  END 111.2  104.5.  PROP 1
STRUCT CABL BEG 112.4 103.  END 112.4  104.5.  PROP 1
STRUCT CABL BEG 113.6 103.  END 113.6  104.5.  PROP 1
***********Gallery 4
STRUCT CABL BEG 136.2 103.  END 136.2  104.5.  PROP 1
STRUCT CABL BEG 137.4 103.  END 137.4  104.5.  PROP 1
STRUCT CABL BEG 138.6 103.  END 138.6  104.5.  PROP 1
*************************** Split 1
*sTRUCT CABL BEG 73.45 103.  END 73.45  104.5. S 5   PROP 1
*STRUCT CABL BEG 74.7  103.  END 74.7   104.5. S 5    PROP 1
*STRUCT CABL BEG 75.95 103.  END 75.95  104.5. S 5  PROP 1
*************************** Split 2
*STRUCT CABL BEG 98.45 103.  END 98.45  104.5. S 5            PROP 1
*STRUCT CABL BEG 99.7  103.  END 99.7   104.5. S 5            PROP 1
*STRUCT CABL BEG 100.95 103.  END 100.95  104.5. S 5         PROP 1
*************************** Split 3
*STRUCT CABL BEG 123.45 103.  END 123.45  109. S 5           PROP 1
*STRUCT CABL BEG 124.7  103.  END 124.7   109. S 5  PROP 1
*STRUCT CABL BEG 125.95 103.  END 125.95  109. S 5           PROP 1
STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9  A 3.14E-4
****************************************************
*With splitting of 2 pillars Save as ncsplit.sav
****************************************************
RET
```